

CHAPTER - I

MINERAL PROCESSING

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1.1 INTRODUCTION

The minerals and the metals are integrally associated with the development of human culture and growth of the society. Minerals are essential component of Nation's material and economic base. They contribute to industrialization and form an important infrastructure for the development of the country. However, conservation of these non-renewable assets is absolutely essential in view of the rapid industrialization and consumption of minerals. Attempts should be aimed to use the resources wisely and efficiently for complete utilization of the components with no waste. Ignoring the awareness in respect of mineral resources, environmental issues have come into prominence and the aspect of conservation and safe guarding the environment can be managed by making proper use of the mined materials, wasting the least, reclamation, restoration, afforestation and creation of better environment for better habitat.

Resource wise, India is well endowed with large variety of rich mineral resources amounting 120 million tonnes in 2002. India produced as many as 64 minerals, constituting 4 fuel minerals, 10 metallic minerals and 50 non-metallic (industrial minerals) in 2003-04. India's ranking in 2002-03 in world production was 2nd in barytes, chromite and talc/steatite/pyrophyllite, 3rd in coal & lignite, 4th in iron ore and kyanite/sillimanite/andalusite, 6th in bauxite and 7th in manganese ore. However, with rapid expansion of mineral based industries, high grade ores are preferentially being consumed resulting with fast depletion of the same and the country is gradually leading to depend upon low grade and complex ores.

Mineral beneficiation is a process to separate the economic minerals from the waste rock gangue. The difference of properties between the value minerals and the undesired minerals determine the method of separation. This aspect of separation of minerals and upgrading the useful values suitable for down stream processing, assume greater significance in the context of low grade and complex ore deposits. Recovery of metal values from these resources of low tenor is associated with generation of more and more quantities of waste and greater load on efficient management of these wastes and protection of environment. The mineral engineers are thus confronted with the work of finding efficient, innovative and cost effective process and equipment for solving these problems faced by mineral industries.

Mineral processing involves a number of unit operations starting with the run of mine (ROM) ore as the initial raw material. Traditionally the scope of mineral processing is limited to liberation of individual mineral phases and their separation from each other without altering the identity of any mineral constituents. Mineral processing is the field of study that starts after mining and ends before extraction of metals for metallic ores. Before separation of the individual minerals, an intimate knowledge of the mineralogical composition of the ore, their mode of association, complexity, texture and size are essential to know the optimum size of liberation. The study of mineralogy helps processing personnel in acquiring this knowledge. The liberation of valuable minerals from the gangue is accomplished by comminution i.e., the process of size reduction to the desired size. Actual separation is then effected by utilizing the difference in physical and other properties of value minerals and the gangue viz., size, shape, color, specific gravity, magnetic property, electrical conductivity, radioactivity, specific surface property like affinity or repulsion towards water etc. Technology is accordingly developed to create the suitable process conditions for separation. Often the surface properties are modified to create a greater difference of a specific property for the value mineral and gangue.

For low grade and complex ores, the mineral processing steps are generally carried out in 'wet' condition i.e. using water as a medium. This is obviously for a number of reasons either demanded by the process including requirement of liberation or for ease and efficiency of handling & separation or other aspects. Whatever be the reasons, the concentrates produced in fine sizes and in pulp condition need to be dewatered, dried and again agglomerated into a suitable lump form for ease of handling for subsequent treatment for metal extraction.

With the above introduction, somewhat details of mineralogical aspects, beneficiation, de-watering and agglomeration are presented in the following pages.

1.2 Importance of Mineralogy in Mineral Processing

Mineral processing technology is evolved to separate and recover ore minerals from gangue in a commercially viable method and is mainly based on the process of mineral liberation and the process of mineral separation. Though ore is mined with the highest efficiency of the technology, the excavated ore gets partly contaminated by the surrounding host rock (overburden) and the geological material closely associated with the ore during mining. Both the materials are undesirable and hence form the gangue. Thus, the contributors to gangue may be igneous rock, sedimentary rock (shale, quartzite, phyllite), hydrothermal vein material, quartz reef and their

metamorphic equivalents. For example, chromite and magnetite may form in a magmatic crystallization process whereas iron ore may form in a sedimentary environment with the episodes of metamorphism, supergene enrichment and martitisation. So, the former is associated with ultramafics (dunite, peridotite, serpentinite), basalt, gabbro whereas the latter is associated with shale, jasper, quartzite, and their weathered equivalents (kaolinitic and gibbsitic shale, quartzite, laterite etc.). So, the expected gangue minerals are different in these two cases. Similarly, the sulphide mineralisation is associated with hydrothermal veins (quartz vein or quartz reef) intruded into various rock types (granites, basalt, meta-basalt, meta-sedimentary rocks). The Table 1 enlists the possible ore minerals and gangue minerals in the ores from wide range of deposits.

From mineral processing point of view, it is important to identify the ore and gangue minerals (mineralogical study), their textural relationship (grain size, grain boundary relationship etc.). The mineralogical study also helps in the modal distribution of ore and gangue minerals which decides the grade of ore. Grain size of the minerals and their textural relationship helps in deciding the size reduction and liberation in mineral processing. An effective liberation of gangue minerals and ore minerals influences the optimum separation efficiency. .

Mineralogical study may be described under the following heads.

1. Mineral identification by optical microscopy, XRD, SEM, EPMA:

For mineralogical characterization of the ores/rocks, optical microscopy is an invaluable tool. For this, polished sections of ores and thin sections of rocks are prepared. The ores/ore forming minerals are studied under reflected light microscopy while the rocks/ rock forming minerals are studied under transmitted light microscopy. The minerals are identified basically from their optical properties. The optical properties differ from mineral to mineral because their crystal structure is different and the chemical constituents of the mineral; also their relative positions in the crystal lattice sites differ. But mineral kingdom is so vast sometimes it becomes difficult to identify all the minerals using only the optical microscope because there are cases where the optical properties of one mineral very closely resemble another. In such cases the help of advanced characterization techniques such as X-ray diffraction (XRD), Scanning Electron Microscopy (SEM) with micro-chemical analysis by WDS or EDS, and Electron Probe Micro-analysis (EPMA) is taken. The latter techniques help in confirming the mineral phases which is very important. On the other hand, the textures and liberation characteristics are studied under optical microscopes.

Mineral identification by X-ray diffraction technique is based on the diffraction peaks at various 2θ values corresponding to d-spacing characteristic to the mineral. As each mineral has a characteristic crystallographic identity, its d-spacings are characteristic and the set of diffraction peaks are characteristic based on Bragg's law. This technique is useful for identifying clay minerals. In the figure-1, the phases in the iron ore sample are identified as goethite, kaolinite, hematite/martite, magnetite and braunite. The limitation of the technique is for the minerals whose volume percentage is less than 3% in the sample.

Some of the trace phases of very small size can not be studied by optical microscopy or XRD. In that case, SEM and EPMA provide a high resolution image of very high magnification and the chemical analysis is done by Energy dispersive X-ray spectrometry (EDX) or by wavelength dispersive X-ray spectrometry (WDX). By chemical calculations, the empirical formula is computed out and the mineral is identified.

Knowledge of the sizes and intergrowth relationships of ore mineral grains is of great importance. Only through careful examination of the ores in polished section can the optimum grain size for effective liberation be determined. Insufficient grinding may result in loss of valuable minerals in the tailings; over grinding wastes energy and may produce slimes that are difficult to treat. The nature of the boundaries between intergrown particles will show whether or not the rupturing of larger particles during grinding is likely to occur at grain boundaries. Information regarding fractures and fissures, and porosity in the ore minerals is derived from the microscopic study. In figure-2 the mineralogy and texture in various components of iron ore are shown to provide an apparent perception of granular martite.

2. Study of Modal distribution : Modal distribution of minerals by microscopy with image analysis system or by grain counting helps in the statistical distribution of ore and gangue minerals which is directly related to the grade of ore. Such study also helps in the liberation studies. It can also be extended to beneficiation products to find the efficiency of beneficiation

3. Liberation study by microscopy : The liberation of ore and gangue is achieved by size reduction (through grinding). The statistical count of interlocked grains, 'ore minerals free of gangue' (liberated ore) and 'gangue minerals free of ore minerals' (liberated gangue) gives the quantitative percentage of liberation. This data is useful in deciding the extent of liberation required for a desired economy of comminution and beneficiation.

$$\text{Liberation (\%)} = \frac{100 \times \{(\text{No. of free ore minerals}) + (\text{No. of free gangue minerals})\}}{(\text{No. of free ore minerals} + \text{free gangue minerals} + \text{interlocked minerals})}$$

Table 1: Associated ore and gangue minerals in various ore types.

Ore-type Iron ore (Sedimentary and supergene enrichment)	Ore minerals Martite, Hematite Fe_2O_3 , magnetite Fe_3O_4 goethite $\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$	Gangue minerals Quartz, jasper, kaoline, gibbsite, shale
Bauxite	Gibbsite, diaspore, boehmite	Goethite, kaoline, Ilmenite, zircon, anatase
Manganese (i) Sedimentary association (ii) Metamorphic association	Psilomelane, pyrolusite, cryptomelane, manganite, wad Braunite, bixybite,, hollandite, jacobsite, hausmanite, wad	Quartz, feldspar, apatite, clay, micaceous mineral, pyroxene, amphibole, and barite, garnet (?), hematite, hydrated iron oxide (goethite)
Limestone Beach sand	Calcite, aragonite, dolomite Ilmenite FeTiO_3 , zircon ZrO_2 , garnet $(\text{Ca}, \text{Mg}, \text{Fe})(\text{Al}, \text{Fe}, \text{Cr})(\text{SiO}_4)_3$, monazite $(\text{Ce}, \text{Th}, \text{Y})\text{PO}_4$, xenotime TiO_2 etc. Sillimanite (?)	Quartz, chlorite, shale Quartz, amphibole, pyroxene, olivine, tourmaline
Copper (hydrothermal and porphyry type deposit)	Sulphides [chalcopyrite CuFeS_2 , chalcocite Cu_2S , covellite Cu Bornite Cu_5FeS_4] Oxides [cuprite, Cu_2O] Carbonate [malachite $\text{Cu}_2\text{CO}_3(\text{OH})_2$, Azurite $\text{Cu}_3(\text{CO}_3)_2(\text{OH})_2$] Trace: U, Mo, Au	Minerals in phyllite, schist, quartz schist, garnet-chlorite schist, chlorite, biotite, dolomite, shale, chert, quartz, feldspar
Lead-zinc (hydrothermal deposit)	Sulphides (galena PbS , sphalerite ZnS) Carbonate- Cerrusite PbCO_3 Sulphate- Anglesite ZnSO_4 Hemimorphite Trace: Ag, Au, Cd, In, Cu, Sb	Phyllite, schist, quartz schist, garnet-chlorite schist, chlorite, biotite, dolomite, shale, chert, quartz, feldspar, Barite, Pyrite, pyrrhotite, arsenopyrite, chalcopyrite, argentite
Chromite (magmatic deposit)	Chromitite FeCr_2O_4 , Trace: Ni	Magnetite, dunite (olivine/ serpentine), pyroxene, gabbro (plagioclase, olivine, pyroxene)
Gold	Arseno-pyrite, native gold	Minerals of host rock (Phyllite, schist, quartz schist, amphibole, chlorite, biotite, dolomite, shale, chert, quartz, feldspar), pyrite, chalcopyrite

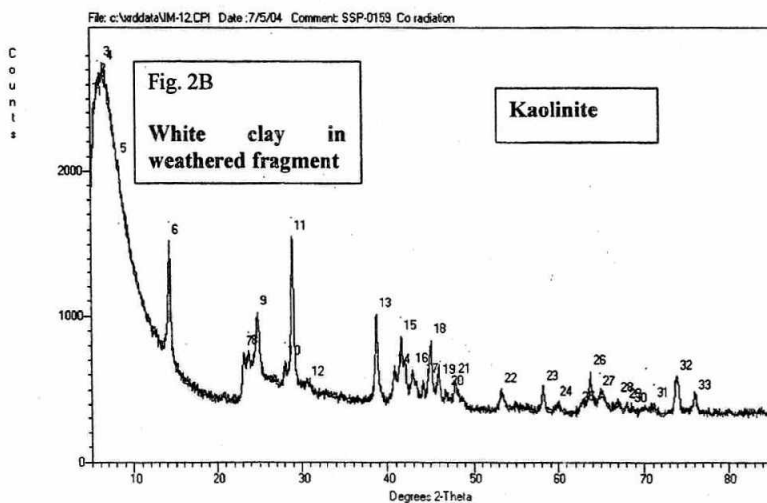
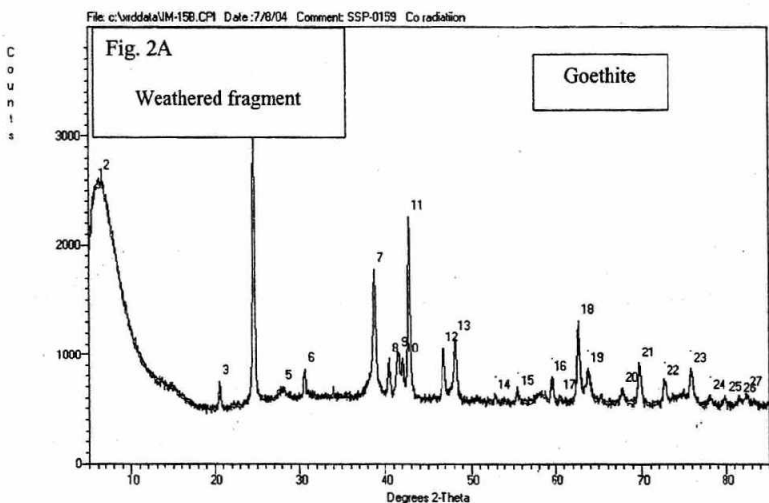


Fig. 2 : X-Ray diffractograms of selected ore fragments from the bulk iron ore sample.

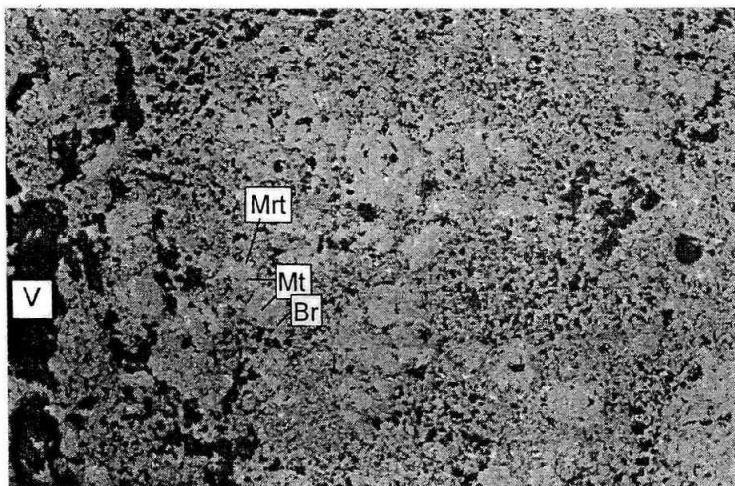


Fig. 3A : Microphotograph of partly martitised magnetite within a matrix of braunite (Br). The martitised magnetite have a core of magnetite (Mt) with an outer rim of martite (Mrt). Plane polarised light X200. The longer edge of photo equals to 0.6mm.

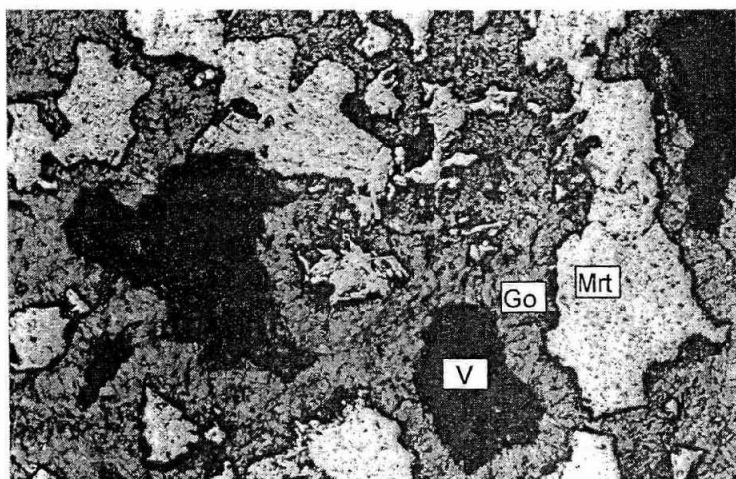


Fig.3B : Microphotograph of martite (Mrt) and inconspicuously developed microplaty hematite around which goethite (Go) is deposited in the voids (V) by precipitation from solutions mobilised. Plane polarised light X200. The longer edge of photo equals to 0.6mm.

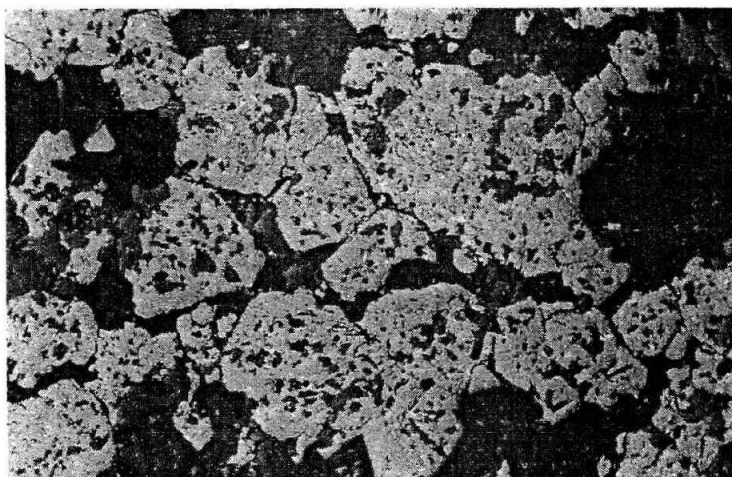


Fig. 3C : Microphotograph of martite (with high reflectivity) and quartz (grey colour with low reflectivity) association and void. Plane polarised light X200. The longer edge of photo equals to 0.6mm.

1.3 RELEVANCE OF SIZE DISTRIBUTION

Valuable mineral matter is usually finely disseminated in the bulk ore containing large proportion of undesirable gangue material. Mineral Processing aims at enriching the ore by getting rid of some of this gangue material making use of differences in many of the physical properties of the two. In order to be able to achieve this, the valuable mineral must be physically liberated from the gangue by size reduction as indicated by the liberation characteristics. In doing so, we end up having a wide range of sizes of the ground ore. This range and the material content in each smaller size ranges are collectively known as size distribution.

This distribution is a very important aspect in mineral processing. There are several issues associated with it. The most important is the fact that size reduction or comminution is an energy intensive operation. Therefore, the final size distribution must not be finer than what is required for liberation. Also, the size distribution should be such that the amount of ultrafines in the distribution is minimum. This is necessary for the fact that processing fines is extremely difficult and expensive.

Due to the nature of the ores the valuable mineral or the gangue may get segregated in certain size classes. In such cases downstream processing may take advantage of such phenomena. Hence, the importance of size distribution is widely acknowledged.

The following example elaborates the utility of size distribution. In this example 1 ton of boulders have been broken into particles less than 1 mm. The crushed material is sieve analysed and the data are shown below in Table 2.

Table 2 : Typical size distribution data

Size range (micron)	Weight (kg)	Weight (%)	Passing Size (microns)	Passing (%)
1000-500	30.8	3.08	1000	100.0
500-250	209.4	20.94	500	96.92
250-150	256.8	25.68	250	75.98
150-75	212.3	21.23	150	50.30
75-38	163.6	16.36	75	29.07
38-0	127.1	12.71	38	12.71

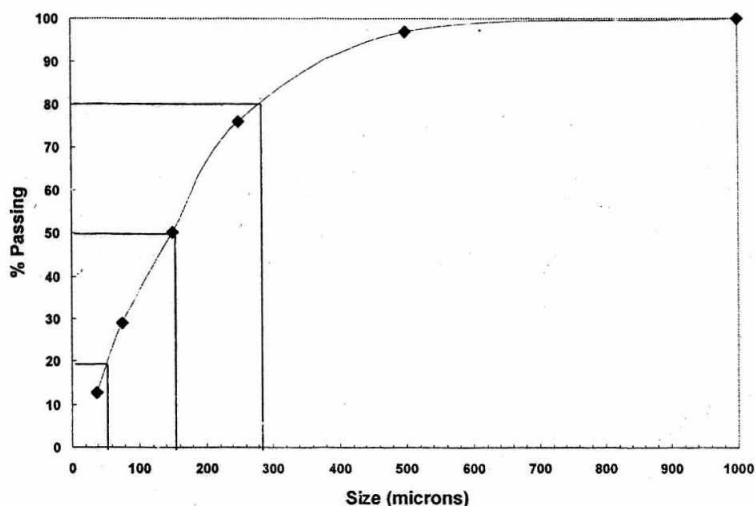


Fig. 4 : Size distribution plot

Usually, the data in the last two columns are plotted with size in the abscissa as shown in Figure 4. In the Figure, three parameters are shown. These are the sizes at which 20%, 50% and 80% of the material pass. These sizes are known as d_{20} , d_{50} and d_{80} respectively and are used to quantify the distribution. In this example, they are 52, 148 and 282 microns respectively.

There are several mathematical functions to represent various size distributions. The most common one is known as the Rosin-Rammler distribution. According to this distribution, if the passing size is denoted as d_p and the percentage of material passing this size as $P(d_p)$ then

$$P(d_p) = 1 - \exp[-(d_p / d_p^*)^\alpha] \dots\dots\dots (1)$$

Where d_p^* and α are characteristic parameters of the size distribution.

1.4 COMMINATION

The process of size reduction is known as comminution. In mineral processing parlance, comminution in coarse range is known as crushing and in fine range it is called grinding.

Generally, crushing is carried out in two stages, namely, primary and secondary crushing. Jaw and Gyratory type crushers are used mostly for primary crushing. They are characterized by wide input side (known as gape) and narrow discharge. They can handle large tonnage of material. Jaw crushers produce a reduction ratio of 4:1 to 9:1 while gyratory crushers produce a somewhat larger range of 3:1 to 10:1. The reduction ratio is defined as the representative feed size by representative product size.

Once the run-of-mine (ROM) ore is crushed to smaller pieces by the primary crushing units, the secondary crushers are employed to achieve further reduction in size. Examples of secondary crushers are cone crusher, impact crusher and roll crusher. Cone crushers produce reduction ratios in the range 5:1 to 8:1. Very high reduction ratios, 20:1 to 40:1, can be achieved using hammer type impact crushers. However, roll crushers do not offer high reduction ratios. They can attain ratios only in the range 2:1 to 4:1. Roll crushers also have the limitation on feed size. Very large size particles can not be gripped by the rollers and the angle of nip is the key parameter in this regard. Rotary breakers are also used for comminution in coarse size range particularly in coal preparation.

Comminution is a very energy intensive operation. Generally speaking, the more energy that is absorbed by the particle, the finer the average size of the product population. If d_p is a representative size of a distribution (usually d_{80}), the relationship between comminution energy absorbed per unit mass and the representative size is defined by a differential equation:

$$dE/dd_p = -kd_p^{-n} \dots\dots\dots (2)$$

where, E is the energy absorbed and n is an exponent whose various values have been suggested by different workers.

The above equation can be solved with the initial condition that $E=0$ when $d_p=d_{pi}$ (product size = parent size) to get (for $n \neq 1$):

$$E = \frac{k}{n-1} \left[\frac{1}{d_p^{n-1}} - \frac{1}{d_{pi}^{n-1}} \right] \dots\dots\dots (3)$$

Thus, when $n = 1$ (Kick's Equation):

$$E = k \ln \frac{d_{pi}}{d_p} \dots\dots\dots (4)$$

When $n = 1.5$ (Bond's Equation):

$$E = 2k \left[\frac{1}{d_p^{1/2}} - \frac{1}{d_{pi}^{1/2}} \right] \dots\dots\dots (5)$$

When $n = 2$ (Rittinger's Equation):

$$E = k \left[\frac{1}{d_p} - \frac{1}{d_{pi}} \right] \dots\dots\dots (6)$$

The later two equations are generally used to estimate the energy consumption for comminution. The work index of a material is calculated using Equation (5), that indicates the ease or difficulty of comminution in terms of energy requirement.

Grinding machines in the mineral industry are of tumbling mill type. These mills exist in a variety of kinds such as ball, rod, pebble, autogeneous, semi-autogeneous, etc. Grinding action is induced by relative motion between the particles of media - the rods, balls or pebbles and the particles themselves. High compression roll mill and fluid energy mills are recent developments in comminution technology.

There are two different types of motion of media particles in the mill, namely, cascading and cataracting generating from the tumbling motion of the mill. When the particles move along the inner surface of the mill shell, lifted up, loses contact with the surface and travel downward in a trajectory through the empty space inside the mill resulting in an impact on contact with the inner surface again, the motion is called cataracting. This motion produces less amount of fines.

When the media particles move up and then roll down along a parabolic path while remaining within the bulk itself, the motion is called cascading. This motion generates fines and to be minimised to the extent possible. Clearly, at lower rpm of the mill cascading is predominant and higher speed is necessary for cataracting motion. However, this is restricted by the critical speed, a very crucial parameter, of the mill.

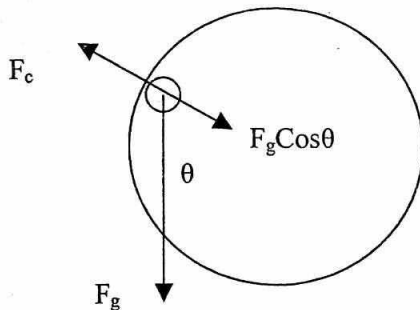


Fig. 5 : Forces on a media particle in a tumbling mill

When a media particle is moved up the two forces acting on it are the centrifugal force F_c and the gravitational force F_g . Balancing them in the radial direction and simplifying,

$$\omega = \left[\frac{2g}{D_m} \right]^{1/2} \dots\dots\dots (7)$$

where ω is the angular speed and D_m is the mill diameter. Expressing angular speed in revolutions per minute,

$$N_c = \frac{42.3}{D_m^{1/2}} \dots\dots\dots (8)$$

This is the critical speed of the mill beyond which the media particles will remain centrifuged at the wall resulting in no impact or grinding action. Thus, the mill must be operated below the critical speed.

In industrial practice, most comminution operations are closed circuit except primary crushing. A comminution circuit is said to be closed when it operates in series with a size classifier and the coarse fraction of the classifier is re-circulated back into the comminution unit. A secondary crusher with a vibrating screen and a ball mill/rod mill with a hydrocyclone are most common closed circuit comminution operations in mineral processing plant practice.

Most industrial grinding circuits are operated under wet conditions. This circuit ensures a steady output of desired sized particles with a suitable distribution. The mass flow rate of the output of this circuit must remain reasonably constant with a pre-set value of the representative size (d_{80}). Deviation from this target will result in under-utilization or choking of downstream processing stages. There are several variables that can be monitored and controlled to achieve this target. Mill charge, feed rate, pulp density, classifier feed pressure, etc., can be measured and adjusted to the required value to ensure smooth operation and obtain output of required specifications.

1.5 CLASSIFICATION

Classification is a method of separating of minerals into two or more products on the basis of size. In coarse size range this may be achieved under dry conditions. However, starting from grinding most operations are generally performed under wet conditions in mineral processing. When a solid particle falls in fluid (water) medium there is resistance to this movement and the value increases with velocity. When equilibrium is attained between the gravitational and fluid resistances forces, the body reaches its terminal velocity and thereafter falls at a uniform rate. This terminal velocity is function of the particle size and density.

Classifiers consist essentially of sorting column in which a fluid is rising at a uniform rate. Particles introduced into the sorting column either sink or rise according to whether their terminal velocities are greater or less than the upward velocity of the fluid. The sorting column therefore separates the feed into two products - an overflow consisting of particles with terminal velocities less than the velocity of the fluid and an underflow of particles with terminal velocities greater than the rising velocity.

Classifiers are divided mainly into two broad classes depending on the direction of flow of the carrying current. Horizontal current classifiers such as mechanical classifiers, spiral classifiers, rake classifier are essentially of the free settling type. Vertical current or hydraulic classifiers such as elutriators, hydrocyclones, hydrosizers are usually of hindered settling types. Sieve bends are also commonly used for classification.

Hydrocyclone

Hydrocyclones (Figure 6) are continuously operating classifying devices that utilise centrifugal forces to accelerate the settling rate of particles. It is one of the most important devices used in the mineral industry. Hydrocyclones operate under pressure. The feed, a mixture of water and solids, enters the hydrocyclone tangentially through the inlet, which forces the mixture to spin inside the cyclone. This spinning motion generates centrifugal forces, which cause the air to disengage quickly and exit through the vortex finder.

The liquid passes down into the conical section where the reduction in diameter accelerates the fluid thus generating centrifugal forces strong enough to cause the solids to separate from the liquid. The larger particles are forced towards the wall because of greater mass and then travel down the length of the conical section of the hydrocyclone in a spiral pattern towards the solids outlet, termed the underflow. The liquids migrate towards the center of the hydrocyclone where the flow reverses and moves upwards towards the overflow through the vortex finder. The finer particles do not get centrifuged

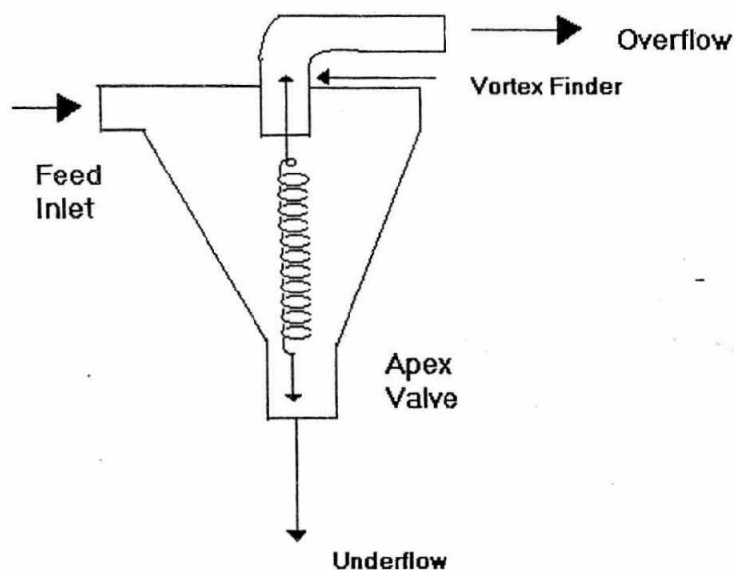


Fig. 6 : Schematic of a hydrocyclone classifier

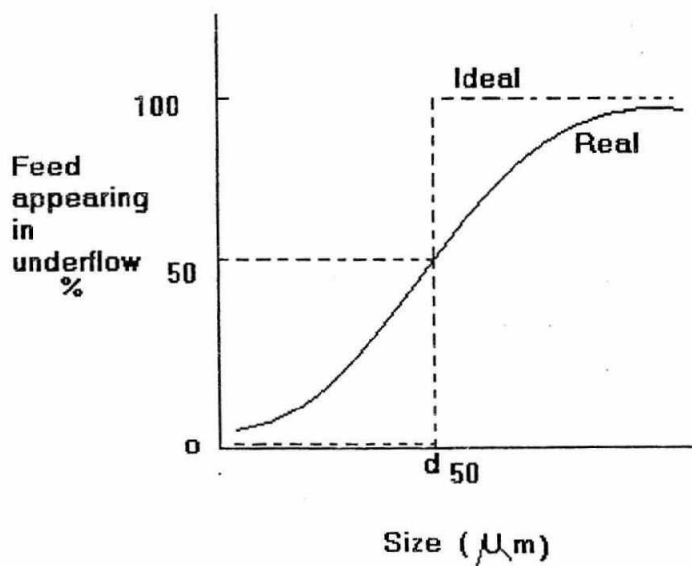


Fig. 7 : Partition curve of a hydrocyclone

towards the periphery due to their smaller mass and hence accompany the liquid to the overflow. Thus a separation of larger size particles from smaller size ones are achieved.

The commonest method of representing cyclone efficiency is by a performance curve or partition curve as shown in the Figure 7. This relates the weight fraction or percentage of material in each size in the feed that reports to the apex or underflow to the particle size. The cut point or separation size of the cyclone is often defined as that point on the partition curve for which 50% of particles in the feed of that size reports to the underflow. Particles of this size have an equal chance of going either with the overflow or underflow. This point is usually referred as the d_{50} . The sharpness of the cut depends on the slope of the central section of the partition curve; the closer to vertical is the slope, the higher is the efficiency. The slope of the curve can be expressed by taking the points at which 75% and 25% of the feed particles report to the underflow. These are the d_{75} and d_{25} sizes, respectively. The efficiency of separation or the so called imperfection I , is then given by

$$I = (d_{75} - d_{25}) / 2d_{50} \dots\dots\dots(9)$$

Several factors affect the performance of a hydrocyclone. The effects of changing operating and design parameters are given below in the Table 3.

Table 3 : Factors affecting hydrocyclone performance

Parameter	Change	Effect	Reason
Vortex Finder diameter	Increase	Decrease Efficiency	More likely to suck particles up
Pressure Drop	Increase	Increase Efficiency	Flow rate increases, Increase in G forces
Apex diameter	Increase	Prevents Overload, And in the extreme case decreases efficiency	Causes flushing of particles and fluid
Feed Flow Rate	Increase	Increase Efficiency	Increase G forces
Vortex Finder length	Increase	Decrease Efficiency	More likely to suck particles up
Cyclone dia.	Increase	Decrease Efficiency	Decrease G forces
Cyclone Length	Increase	Increase Efficiency	Residence time increases
Particle size	Increase	Increase Efficiency	More likely for particle to migrate to exterior wall

There are a number of empirical relationships that are used by designers in predicting performance and designing cyclones. There are different equations to calculate the cut-point d_{50} . Plitt's equation is one of the most important of them:

$$d_{50} = [14.8 D_c^{0.46} D_i^{0.6} D_o^{1.21} \exp(0.063V)] / [D_u^{0.71} h^{0.38} Q^{0.45} (S-L)^{0.5}] \dots\dots(10)$$

Where, d_{50} is the cut point (μm), D_c is the cyclone diameter, D_o is overflow diameter, D_i is inlet diameter, V is the volumetric percentage of solids in feed, D_u is the underflow opening diameter, h is the effective cyclone length, Q is total volume flow rate, S and L are specific gravity of solids and liquid respectively.

1.6 GRAVITY CONCENTRATION

Gravity concentration process that exploits the differences in densities of minerals to bring about a separation is the oldest beneficiation method known to mankind. Although with the advent of froth flotation, the relative importance of gravity concentration has declined in twentieth century, on an average higher tonnage of material is still treated by gravity concentration than flotation. It finds diverse applications in the treatment of coal, beach sands, iron, gold, diamonds platinum, barite, fluorspar, tin, tungsten ores etc. The gravity separation processes are comparatively cheaper and environment friendly.

Gravity separation of two minerals with different specific gravity is carried out by the relative movement in response to force of gravity and one or more other forces. Normally one of the forces is resistance to motion by a fluid, usually water. Besides the specific gravity, the factors such as size, shape and weight of the particle also affect the relative movement and hence, the separation. The ease or difficulty of separation depends upon the relative differences in these factors.

The Concentration Criteria (CC) which gives an idea of the amenability of separation of two minerals, can be expressed by

$$CC = \frac{(d_H - d_F)}{(d_L - d_F)} \dots\dots\dots(11)$$

where, d_H = Sp. gr. of the heavy mineral
 d_F = Sp. gr. of the fluid, and
 d_L = Sp. gr. of the light mineral.

Generally, when the quotient is greater than 2.5 (whether positive or negative) then gravity separation is relatively easy. With a decrease in the

value of the quotient the efficiency of the separation decreases and below 1.25 generally, gravity concentration is not feasible.

As mentioned above, besides the specific gravity, the motion of a particle in fluid also depends on its size. The efficiency of gravity concentration increases with an increase in particle size. The particle movement should be governed by the Newton's Law, Eq. 12

$$v = \left[3gd \frac{(D_s - D_f)}{D_f} \right]^{1/2} \dots\dots\dots(12)$$

where, v = terminal velocity of the particle, D_s = density of the solid, D_f = density of the fluid, and d = diameter of the particle.

For small particle, the movement is dominated mainly by surface friction and these respond poorly to commercial, high capacity gravity separators. To reduce the size effect and for making the relative motion of the particles specific gravity dependent, a closely sized feed is desirable.

There is no single mechanism for the operation of a particular gravity separator. Generally a combination of two or more mechanisms is helpful in explaining the behavior of any separator. The various mechanisms proposed are briefly described below.

Density

The methodology employs a fluid with the apparent density in between those of the minerals to be separated. Hence due to difference in the buoyancy, one mineral floats while the other sinks. The most common example is the heavy medium separation.

Stratification

In this case the minerals are stratified by an intermittent fluidization caused by the pulsation of the fluid in a vertical plane. Examples are various types of jigs used for concentration including Baum and relatively more recent Batac jigs.

Flowing Film

The minerals are separated by the relative movement through a stream of slurry which is flowing down a plane by the action of gravity. Examples are sluice, Richert Cone etc.

In another type of flowing film concentrators, the various constituents are separated by the superposition of a horizontal shear force on the flowing film. Examples are Shaking table, Bartles-Mozley Separator and Cross Belt Concentrator.

Range of the Available Gravity Concentrators

A wide range of gravity separators is available for concentration of various types of ores with feed varying particle size distribution. Besides the cost involved, the important factors in equipment selection are the particle size distribution of the feed, specific duty required, throughput and efficiency of the separation desired.

Recent Developments in Fine Gravity Concentration

As mentioned in the previous section, gravity concentration processes suffer from serious limitations in treating fine particles (typically below 50 micron) efficiently. The factors such as small mass, low momentum, colloidal coating, hetero-aggregation, high surface area, increased surface energy and viscosity cause loss in selectivity of separation. But considering the high loss of values in fines and slimes coupled with the environmental pollution problems, there have been considerable efforts to develop an efficient gravity separator for fines. The various early fine gravity concentrator like Buddles, Stakes, Vanners, Round tables and Round frames were relied upon the principle of thin film concentration and suffer from very low capacity per unit area and the low ratio of enrichment.

In recent times because of their simple design and less maintenance problems, water only cyclones are gaining popularity. The equipment is similar to conventional cyclone except that it has got a large angle lower conical section. This helps in suppressing the classification and leads to separation based on the difference in the specific gravity of the suspended particles. The equipment has been used for coal preparation but there exists scope for extending its application to lead-zinc, cassiterite, placer deposits of gold etc.

The application of centrifugal forces to heavy media separation, in the D.M.S. Cyclone, Dynawhirpool and the Triflow separator, has increased the range of particle sizes that can be treated down to 200 microns. The recent fine gravity centrifugal separators like Knelson concentrator, Kelsey jig and Multi-Gravity separator (MGS) can treat particles further in the finer size range.

1.7 MAGNETIC SEPARATION

This is one of the concentration processes that utilize the differences in magnetic properties of various minerals present in the ore body. The magnetic fraction may be valuable or gangue depending upon its use in a particular flow sheet or process and so also the non-magnetic fraction, e.g., separation of magnetite (magnetic) from quartz (non-magnetic), separation of tin bearing mineral cassiterite (non-magnetic) from magnetite (magnetic) impurity etc.

When a material is placed in a magnetic field H , the magnetic field inside the material will be the sum of the external magnetic field and the magnetic field generated by the material itself. The magnetic field that exists in such a material is called the magnetic induction or magnetic flux density, B , and it is defined as follows:

$$B = \mu_0(H + M) \quad \dots\dots\dots(13)$$

Where, M = intensity of magnetization (a property of the material)
 μ_0 = permeability of free space

The magnetic property of minerals of most interest to mineral processing engineers is the magnetic susceptibility (c) which is defined by $c = M/H$. The most commonly tabulated value of magnetic susceptibility is the specific susceptibility in electromagnetic units. The specific susceptibility (cs) is defined as the ratio of specific magnetization (Ms) to the magnetizing field, $cs = Ms/H$. All minerals are affected in some way when placed in a magnetic field, although the affect is too small to be detected in most of the minerals. Magnetic susceptibility of some common minerals is given in Table -4.

Table 4 : Magnetic susceptibilities of common minerals

Mineral	Specific susceptibility ($10^{-6} \text{ m}^3/\text{kg}$)	Mineral	Specific susceptibility ($10^{-6} \text{ m}^3/\text{kg}$)
Calcite	-0.3 to -1.4 0	Pyrrhotite	10 to 30,000
Quartz, Feldspar, Magnesit	-0.5 to -0.6	Hematite	10 to 760
Kaolinite	-2	Ilmenite	46 to 80,000
Halite, Gypsum, Anhydride	-0.5 to -2.0	Magnetite	20,000 to 110,000
Serpentinite	120 to 290	Dolomite	-1 to -41
Illite, Montmorillonite	5 to 13	Sandstones, Shales, Limestone	0 to 1200
Biotite	5 to 52	Serpentine	110 to 630
Goethite	26 to 280	Clay	10 to 15
Chalchopyrite	0.6 to 10	Coal	1.9
Pyrite	1 to 100		

In general, minerals can be classified in two broad categories, according to whether they are attracted or repelled by a magnetic field.

Diamagnetic: Diamagnetic minerals are made up exclusively of diamagnetic ions and are repelled along the lines of magnetic forces to a point where the field intensity is smaller. Diamagnetic ions have no unpaired electrons. All their d- and f- electrons are in closed shells. The forces involved here are very small and diamagnetic minerals cannot be concentrated magnetically.

Paramagnetic: Paramagnetism in minerals generally arises from ions with unpaired electron spins, most commonly of the first transition series.

They are attracted along the lines of magnetic force to points of greater field intensity. Paramagnetic minerals can be concentrated in high intensity magnetic separators.

Ferromagnetism: Ferromagnetism is a special case of paramagnetism, involving very high forces. In a few minerals, notably Fe, Ni, Co, Mn, Cr, Ce, Ti and Pt group metals, the interaction between the spins cause spins on adjacent atoms in the minerals to become aligned parallel to each other. This suggests that all unpaired spins may become parallel and that a permanent magnetization may exist even in the absence of external magnetic field (remanence). They can be separated in a low intensity magnetic separator.

Fundamentals of magnetic concentration

Concentration is achieved by simultaneously applying to all particles in an ore a magnetic force that acts on magnetic particles and a second force or combination of forces which acts in a different direction and affects both magnetic and non-magnetic particles. The most commonly applied nonmagnetic forces are gravitational, centrifugal and fluid drag. Other forces that usually enter in an incidental manner are frictional, electrostatic, Van Der Waals, and capillary. A magnetic separator is generally classified as low intensity if its maximum field intensity is less than about 2000 gauss ($H = 1.6 \times 10^5 \text{ A/m}$, $B = 0.20 \text{ T}$). Low intensity magnetic separators (LIMS) are used to treat ferromagnetic and highly paramagnetic minerals such as iron and magnetite. High intensity magnetic separators (HIMS) generally have field strengths of 10 to 20 kilogauss. These separators are used to treat weakly magnetic minerals, such as hematite.

Magnetic separators are commonly classified into two broad groups, namely, wet and dry based on their usage. A more definitive classification within these two basic groups is made based on the relative magnetic field strength of the individual units, i.e. wet low intensity magnetic separators, wet high intensity magnetic separators etc. Table-5 shows the basic groupings of the most commonly used magnetic separators.

Table 5 : Basic Groupings of Common Magnetic Separator

Wet magnetic separators	Dry magnetic separators
<p>Low : Drum separators Bowl traps Magnetizing coils and blocks Demagnetizing coils</p> <p>High : High intensity separators</p>	<p>Low : Magnetic pulleys Rectangular suspended magnets Magnetic drums-radial pole types Magnetic drums-axial pole types Plate magnets Grate magnets</p> <p>High : Induced roll magnetic separators Cross belt magnetic separators Ring type magnetic separators</p>

Common Types of Magnetic Separators Used for Concentration

Ore Cobbing Magnetic Pulleys : Ore cobbing or concentrating magnetic pulleys utilize more poles across the pulley width so as to develop as uniform field depth as possible and a sufficient area of collecting magnetic poles to carry the large amount of magnetic material commonly encountered in such applications. Generally, ferromagnetic minerals are used for concentration.

Magnetic Drums : Magnetic drums with axial pole design are used to concentrate ferromagnetic minerals. Feed materials up to 1-inch diameter can be treated. The drum speed can be varied between 20-45 rpm in low intensity whereas it is up to 200 rpm for high intensity separators.

Induced Roll Magnetic Separator : It develops high intensity magnetic fields and is capable of removing particles that do not respond to the low intensity magnetic separator. This is widely used to treat beach sands, wolframite, tin ores, glass sands and phosphate rocks.

Cross-belt High Intensity Magnetic Separators : A cross belt runs across the face of the electromagnetic pole, and the sharp magnetized points of this upper pole attract the weakly magnetic material. The cross belt transports it to a suitable discharge point. Selective mineral concentration of weakly magnetic minerals e.g. ilmenite, monazite, garnet, chromite, wolframite, etc., can be separated by using this instrument.

Ring Type Magnetic Separators : The basic construction is similar to the cross belt but a magnetized steel ring is substituted for the cross belt.

Low Intensity Wet Drum Magnetic Separators : This is used to concentrate ferromagnetic particles such as iron of abrasion, magnetite and some pyrrhotites. The feed size is limited to 1/8 inch or even finer. Two well known usage are the concentration of magnetic taconite ores and the recovery of magnetite media in heavy media separation plants.

High Intensity Wet Magnetic Separators : High intensity separators generally use a field strength of about 20,000 gauss. Use of a matrix of shaped iron pieces which produce high field gradients to act as collection sites for paramagnetic particles. The commonly used matrix to form the high gradient sites is balls, rods, grooved plates, expanded metal and fibers. High gradient magnetic separators use uniform field of a solenoid. The core is filled with a matrix of secondary poles such as ball bearings or wire wool to obtain the high gradient.

Super Conducting Separators : Small laboratory super conducting solenoids with fields up to about 60 kilogauss are commonly available and are used for the production of large volumes of relatively permanent magnetic field. Negligible power loss is an important advantage.

1.8 ELECTROSTATIC SEPARATION

Electrostatic separation is one of those important unit operations where electrical conductivity property of mineral surface is used selectively to separate out desirable mineral from other undesirable minerals. Electrostatic forces are generated by the action of an electric field on a charged particle. Consequently, in any electrostatic separation process one needs a source of electrical potential to generate the electric field and a process by which the individual particles are charged electrically. It is found that the following factors have significant effect on the process:

1. Intensity of electric field
2. Particle size
3. Relative humidity
4. Temperature of the particle/bed
5. Inter-electrode distance

Electrostatic separation has the following advantages over any other processes:

1. The electrostatic forces work on the particles to be separated only; they do not affect the medium in which the particles are located
2. The trajectories of the particles under the influence of the electric field follow the electric field lines. The electric field lines may be shaped to suit the particular application
3. The direction of electrostatic forces may be reversed by either changing the polarity of the charge or the direction of the external electrostatic field
4. The electrostatic forces may be arranged to work in combination with other forces such as gravitational or centrifugal forces
5. The electrostatic separation forces are independent of the substrate of the material on which the surface electric charge is generated. They are determined solely by the product of electric field and charge. In magnetic separators the forces are considerably greater, yet such forces work on magnetic materials only. The electrostatic forces do not differentiate between magnetic and non-magnetic materials. The charged magnetic particle placed in an electric field will be subjected to forces practically equal to those acting on a similar particle made out of non-magnetic material and charged with the same charge.

However, the process is associated with the following disadvantages as well:

1. Limitation of maximum mass that it can effectively work upon.
2. The size of the material to be separated should be very small which leads to the increase of comminution cost.

Electrification Process

The separation by this process can be achieved by selective charging of the particles which is known as electrification. Different electrification processes are utilised in practice. They are Tribo-electrification, Corona electrification and Induction electrification.

Electrostatic separation Equipment

Generally two types of equipment are used for separation. These are as follows:

1. Drum type electrostatic separation unit

This equipment consists of a rotating drum made of mild steel or some other conducting material, which is earthed through its support bearings as shown below in the Figure 8. An electrode assembly, comprising of a brass tube in front of which is supported a length of fine wire, spans the complete length of the roll, and is supplied with a fully rectified DC supply of up to 50 kV, usually of negative polarity. The voltage supplied to the assembly should be such that ionisation of the air takes place. This can often be seen as a visible corona discharge. Arcing between the electrode and the roll must be avoided as it destroys the ionisation. When ionisation occurs, the minerals receive a spray discharge of electricity, which gives the poor conductors a high surface charge, causing them to be attracted to and pinned to the rotor surface. The particles of relatively high conductivity do not become charged as rapidly since the charge rapidly dissipates through the particles to the earthed rotor. These particles of higher conductivity follow a path, when leaving the rotor, approximating to the one, which they would assume if there were no charging effect at all.

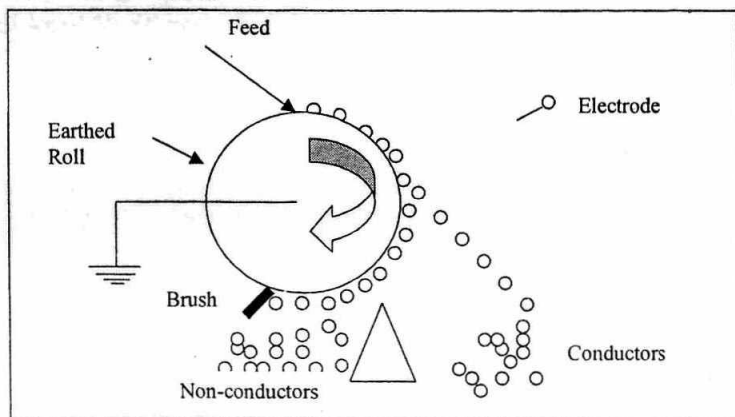


Fig. 8 : Principle of Electrostatic Separation

A combination of pinning and lifting effects can be created by using a static electrode large enough to preclude corona discharge, following the electrode. The conducting particles, which are flung from the rotor, are attracted to this static electrode and the compound process produces a very wide and distinct separation between the conducting and non-conducting particles.

2. Plate type electrostatic separation unit

A plate or screen type electrostatic separator is also used for separation. This type of equipment mainly consists of an oval type, high voltage electrode, which induces the electric field. The material is fed through a sloping, grounded plate under gravity. The electrostatic field is effectively shorted through the conducting particles, which are lifted towards the charged electrode in order to decrease the energy of the system.

Application

Electrostatic separation is used successfully for beneficiation of wide range of minerals. Important examples are:

1. Beneficiation of coal.
2. Beach sand beneficiation.

1.9 FLOTATION

Froth flotation is a process used to separate minerals, suspended in liquids, by selectively attaching them to gas bubbles. Hence, in flotation we have a three-phase system. The separation principle is based on the lack of affinity of the mineral surface towards water, a property denoted by hydrophobicity. Flotation is undoubtedly the most important and versatile mineral processing technique and applications are being expanded to treat greater tonnages and to cover new areas. It is a selective process and can be used to achieve separation from complex ores such as lead-zinc, copper-zinc etc.

Principles of flotation:

Froth flotation utilises the differences in physico-chemical surface properties of various minerals. After treatment with reagents, such differences in surface properties between the minerals within the flotation pulp become apparent. For flotation to take place, an air bubble must be able to attach itself to a particle and lift it to the water surface. The process can be applied to relatively fine particles. In flotation concentration, the mineral is usually transferred to the froth leaving the gangue in the tailing in direct flotation

and in the reverse flotation the gangue is separated into the float fraction leaving the concentrate in the pulp.

In flotation, the minerals can be classified as hydrophobic, i.e., having no affinity towards water and hydrophilic, i.e., having strong affinity towards water. Very few minerals are naturally hydrophobic and the hydrophobic conditions could be achieved by using chemical reagents. Diamonds are naturally hydrophobic and this property is made use of in grease tabling to recover diamonds.

The tensile forces acting on the surface of the particles are responsible for the development of an angle between the mineral surface and bubble surface and this angle is known as contact angle. The flotability of mineral increases with the contact angle. Surfaces with greater contact angles are said to be hydrophobic or aerophilic. Most of the minerals are not water repellent in their natural state and flotation reagents must be added to the pulp to attain selective hydrophobicity of the surfaces. The most important reagents in this regard are collectors. Various other reagents are also used in flotation and are briefly discussed below.

Reagents for flotation process

The reagents employed in flotation are generally interfacial surface tension modifiers, surface chemistry modifiers, and/or flocculants. Usually these are classified as collectors, frothers and modulating agents.

Collectors are reagents that are absorbed on the mineral surfaces to render water repellent property of the surface. The collectors are mainly classified as ionising and non-ionising. The ionising collectors may be divided into anionic and cationic types. Anionic collectors are characterised by an organic-acid group and are the most important type. The cationic collectors are characterised by the cation. Usually, very small quantities of collectors are used in flotation ranging from 0.2 to about 1.0 kg per ton of material processed.

Frothers are another group of reagents used to obtain the stability of the froth. Frothers are similar to ionine collectors. Generally, they are heteropolar, surface-active organic reagents capable of being adsorbed on the air-water interface and reduce the surface tension to form a stable air-bubble. Chemically, the frothers may contain any of the groups such as hydroxyl, carboxyl, carbonyl, amino group and sulpho group. The alcohols containing hydroxyl group are most common. Pine oil, cresylic acid, MIBC are widely used frothers.

Modulating agents are chemical additives that modify the environment of the flotation and may be classified into activators, depressants, pH regulators and flocculants.

Activators are chemical compounds, which interact at the mineral surface thus altering its chemical nature to promote its interaction with the collector.

Depressants are chemical compounds, which again alter the mineral surface to prevent or hinder the action of collectors. They are required to depress certain minerals to promote the selective flotation of desired minerals.

pH regulators are used to control the selective separation of the minerals and can be achieved by using a variety of bases and acids.

Flocculants are another class of reagents that may effectively control flotation in certain cases. They are essentially surfactants like collectors but having different effect and hence different application. Their principal characteristic is a polymeric hydrocarbon chain, which bridges together large number of fine particles producing an aggregate of them called flocs. Unlike a single fine particle, a floc may have enough mass for effective flotation and help in recovery of fines through the flotation route.

Flotation Mechanism

The process of flotation is a sequence of several operations, some of which are done in the preparatory stage and some in the flotation cell itself. The sequences are:

- 1) Liberation of desired material
- 2) Absorption of reagents on particle surfaces through conditioning
- 3) Generation of air bubbles
- 4) Collision of particles with bubbles and their adsorption
- 5) Generation of air-water mineral complex in the presence of reagents and
- 6) Transport of air-water-mineral aggregate to the surface.

Flotation Machines:

Preliminary laboratory test work on few grams could be conducted in Hallimond tube. In this tube the mineral particles are held on a support of sintered glass inside it containing the distilled water and the collector under test. Air bubbles are introduced through the sinter and hydrophobic mineral particles are lifted by the bubbles. The bubbles burst at the water surface allowing the particles to fall into the collecting tube. Commercial flotation machines may be divided in to mechanical cell, pneumatic cell and cyclone cell.

A mechanical cell equipped with a stator and a rotor to keep the mineral particles in suspension and to disperse air supplied (partly by suction and partly by compression) through a central pipe around the shaft for the rotor.

The stator may be attached to the air pipe or to the cell walls. Manufacturers of mechanical flotation cells in the industry include Denver equipment, Galigher, Wemco, Outokumpu and Sala. Denver type flotation machines are also being manufactured indigenously in India.

In a pneumatic cell, suspension of solid particles in water is achieved by the compressed air being suitably dispersed throughout the volume of the cell. It employs a perforated grid (or pipes) arranged in an appropriate position near the top of the cell. This arrangement allows a thick bed of froth to be formed. The flotation pulp, appropriately prepared and ready for separation, is fed with a minimum of agitation on top of this bed of froth.

A cyclone cell, into which the feed is delivered (together with air) through a cyclone feeder, under pressure. These are known as air-sparged hydrocyclones. Very few of this type are presently in use.

Operating guidelines in flotation machine

1. Peripheral speed affecting amount of air drawn.
2. Pulp density affecting volume of air and power consumption.
3. Pulp feed point below impeller may create choking.
4. Pulp level affects turbulence and aeration.
5. Frother quantity may affect bubble size and air intake.
6. Impeller and stator position affects air intake and specific power consumption.
7. Mechanism of removal of froth affects the quality of froth.
8. Improper removal of chamber product affects efficiency and quality.

Oleo flotation and oil agglomeration are also prevalent in coal industry. Electro-flotation is a recent development in the recovery of ultra fines. Scope of electro-flotation has also been extended to the field of effluent treatment.

1.10 DE-WATERING

The removal of water from solid, either partially or completely, comes under the unit operation, 'Solid-Liquid Separation' which forms an integral part of several mineral processing and hydro-metallurgical processes. In recent years the importance of solid-liquid separation techniques have gained considerable importance in order to reduce the load on thermal dryers which are highly energy intensive and also for environmental protection. The economics of coal utilisation, large scale mineral processing and water treatment operations are critically dependent on the solid-liquid separation technology since any improvement in product moisture reduction techniques can result in significant cost saving.

The term, 'De-watering' in solid-liquid separation is used to define two different operations:

- a) The reduction of slurry volume and production of semi-dried solid which are most commonly achieved through sedimentation. In this case separation is affected by exploiting earth's gravitational field (thickener) or centrifugal field (centrifuge or hydro-cyclone).
- b) The slurry is subsequently de-watered through filtration in which separation of solids from liquid is achieved by passing the slurry through a filtering medium on which solids build up forming cake.

Various models, such as those of Richardson & Zaki, Michaels & Bolger, Rober as well as Kynch etc. are described in the literature to understand the sedimentation kinetics. Basically the whole of settling cycle can be divided into two zones: (a) constant (or maximum) settling rate, and (b) falling rate period.

To understand the phenomenon of de-watering, it is important to define how the moisture could be associated with the solid system. Usually the moisture retention property of a solid system is influenced by the inter-particle geometry, inherent physical flaws in the solid structure, and porosity of the solid itself. For an ideal spherical particulate network moisture could be present in three distinct forms: (i) capillary, (ii) funicular, and (iii) pendular. In filtration process capillary water is partially removed, while pendular and surface water are removed by centrifugation or thermal drying. The process of filtration is often explained in terms of capillary theory which treats the pore system in a filter cake as a bundle of capillaries of changing diameters. The displacement of liquid phase will be in a tortuous way along the capillaries: the capillary with the widest diameter will de-water first after overcoming a certain pressure difference. On further increase of differential pressure the narrower capillaries will de-water subsequently.

The kinetics of de-watering can be interpreted in terms of the theory of flow through porous media and Darcy's law holds good according to which the volume flow rate (U) through a packed bed of unit area is given by:

$$U = (K * \Delta P) / (\mu * L) \dots\dots\dots(14)$$

where,

ΔP is the pressure difference across the filter cake having thickness L

K = permeability of the (cake) bed

μ = the fluid viscosity

The conventionally available de-watering techniques are not sufficient for treating very fine particles (-325 mesh size) due to associated

problems. Hence there is an increasing need to develop improved de-watering techniques. Use of flocculants and surfactants in the de-watering process has been successful in achieving this end to a considerable extent in recent years.

Filtration of fine particles has always been a difficult area of mineral beneficiation as well as coal preparation and has attracted much of R & D effort throughout the world. Traditionally de-watering of fine particles has been carried out using rotary vacuum filters, usually drum or disk filters. During recent years a variety of pressure filters such as (a) air blown filter press and (b) tube press have been developed. Extensive field trials have shown that pressure filter can reduce free moisture down to very low level. However, the widespread adoption of this equipment has been slow because of relatively high capital costs involved. Current R & D is focused on the development of a potentially low cost system, based on the use of an ultra-high speed centrifuge.

R & D effort has been made towards the use of (a) flocculants - chemical (organic) based and more recently bacteria based, and (b) surfactants as de-watering aids. Flocculants are usually linear, long chain, water-soluble polymers based on polyacrylamide. These increase the yield of filtered solid and produce a permeable filter cake amenable to rapid de-watering. In this case, the long chain polymers bridge between individual particles, to produce multi-particle aggregates. Surfactants typically consist of hydrophilic and hydrophobic grouping. Although surfactant enhanced de-watering is of technical and economic importance, the fundamental surface chemical phenomena which decide their mode of action are not fully understood. It is generally believed that these allow filter cake capillaries to drain more rapidly by reducing the surface tension, and thus reduces moisture content of the cake.

1.11 AGGLOMERATION

Often concentrates are produced from mineral processing plants in fine particulate form and as such those can not be utilized for metal extraction unless they are bound into some compacted or lumpy form called 'agglomerates' suitable for handling and feeding into furnaces. The process of 'agglomeration' can be classified mainly into the following groups, namely, Briquetting, Nodulizing (rotary kiln sintering), Vacuum extrusion, Sintering and Pelletizing.

Briquetting : This is a very simple process, in which fine grained materials are pressed into briquettes with the addition of moisture and/or another binder under high mechanical pressure. The briquettes may undergo

further treatment or thermal processing before their use. Although their metallurgical behaviour in melting or reduction furnaces is very good, iron ore briquetting could not make headway since the processing cost is relatively higher and the production capacity of briquetting units is limited compared to sintering or pelletisation units. Also, getting suitable strength becomes a problem and their handling produces undesired fines. The briquetting process is still utilized to agglomerate small quantities of flue dust or other circulating materials produced in integrated iron and steel plants.

Nodulizing : In this process, iron bearing fines and carbon are passed through a rotary kiln, inclined at a few degrees to the horizontal, counter current to hot gases produced by a gas fired burner. The flame temperature is about 1300-1400 °C and the maximum temperature of the agglomerates is about 1100-1200 °C. The exhaust gases leave the kiln at 300-400°C. The hot agglomerate discharges on to a water cooled chute and is removed by a water cooler conveyor. The agglomerate quickly builds up in the walls of the sinter zone to form rings. These are removed at frequent intervals by means of scraper. The product from the kiln is in a semi-plastic state, and when cooled, is dense, slaggy and difficult to reduce.

The ore fines and/or flue dust are mixed with water, and a binder such as bentonite is added if necessary. The mix is fed into a de-airing chamber, which is connected to a vacuum pump, by means of a scroll feeder, and extruded through a series of dies by means of an auger. Short cylinders are produced by means of a knife which slices through the extruded material at regular intervals. The cylinders fall on to a conveyor, where they remain for a curing time of approximately forty-five minutes. They may be charged to the furnace as cured or may be fired.

Pelletisation : This consists of two distinct operations, namely, forming the ball shaped pellets at atmospheric temperature and then firing them at a temperature below the softening temperature. Iron ore fines and concentrates are ground to suitable fineness generally 50-70% below 50 microns and mixed with some quantity of moisture and a suitable binder e.g., bentonite, lime etc and the mixture rolled in either a horizontal drum or an inclined disc. At this stage, the spheres known as green balls or pellets have adequate strength to withstand handling to the firing stage. Pellet firing, called induration, is normally carried out using either a gas or oil. Initially, shaft kilns were used but later, horizontal traveling grates, or a combination of horizontal traveling grates and rotary kilns, were developed for this purpose. During induration, not only the crystal structure is changed but also other bonds appear by reactions between slag forming constituents - both between each other and with iron oxides. In another pelletisation process, called 'Cold bonded

pelletisation', binders like calcium hydroxide or cement is added during green ball formation. They are indurated by using steam under high pressure in an autoclave or keeping the green pellets under moist condition over a long period of time to complete the setting process.

Although, studies on pelletisation started some time in 1913 in Sweden, the first commercial plant came into existence in Sweden in 1950.

Sintering : The sintering process consists, in essence, of mixing iron ore fines, moisture, other fine iron bearing recycling material like mill scale, flue dust etc., fluxing material e.g. lime, limestone, dolomite, quartz etc. and solid fuel, normally coke breeze, and loading the mix on to a permeable grate. The upper surface is ignited by oil or gas burners and air is sucked downwards through the grate. After a short ignition period, heating of the top surface is discontinued and a narrow combustion zone moves downwards through the bed, each layer in turn being heated to 1200-1500°C. In advance of the combustion zone, water is evaporated and volatile compounds are driven off. In the combustion zone, bonding takes place between the grains and a strong agglomerate is formed. Most of the heat in the gases leaving the combustion zone is absorbed by drying, calcining and pre-heating the lower layers in the bed. When the combustion zone has reached the base of the sinter mix the process is complete, and the sinter cake is tripped from the grate and roughly broken up. After screening, the undersize is recycled and the oversize is cooled and sent to the blast furnaces.

The sintering of non-ferrous ore fines is sometimes carried out in the reverse direction. A thin layer of mix is first charged on to the grate and ignited, mix is then added to bring the bed up to the full height and air is blown upwards through the bed. The process is complete when combustion at the bed top has been completed. This process is referred to as up-draught sintering, in contradiction to the normal down draught sintering.

Evaluation of properties of agglomerates : In order to evaluate the suitability of the agglomerated products in the blast furnaces, direct reduction reactors and smelting reduction furnaces, certain test procedures have been assigned by standardization organizations around the world. So far physical properties are concerned, they are strength indices, namely, Shatter Index and Tumbler Index. The other important properties are chemistry, reducibility, reduction degradation index (RDI) and softening-melting characteristics.

Of all the agglomeration processes, only the continuous sintering strand and pelletizing are now of any significance. The reasons for the failure of other processes are:

1. High fuel cost due to thermal inefficiency
2. Mechanical unreliability leading to high maintenance costs and low availability
3. Liability to serious operating problems
4. Unsuitability for large scale production
5. Poor quality of product

Briquetting generally is liable to mechanical problems. Unfired briquettes have very little strength when heated under reducing condition. Therefore, they break down in the furnace stack causing serious dust loss. Fired briquettes are much more satisfactory but the extra cost is a serious economic burden. Vacuum extrusion gives rise to a product which is unlikely to be strong enough to withstand degradation in the blast furnace stack unless it is fired before being charged, which puts up the production cost to an uneconomic level.

Both the continuous sintering and pelletizing process yield a satisfactory product. It is also possible to maintain high plant availability and to scale up the units to a high output rate. The sintering process is particularly suited to treat ore fines, and has the advantage that it is quite practicable to add flue dust and flux to the sinter mix. Coke breeze, which is a by product of the blast-furnace coke manufacturer, is a very suitable fuel. The sintering process is not suitable for agglomerating fine concentrates, and here the pelletizing process is most suitable. When the ore fines or coarse concentrates are to be pelletized, it is usual to carry out further grinding before pelletization is carried out.

1.12 MATERIAL BALANCE, PLANT EVALUATION AND MODELING

Simple mass balance for a processing plant often leads to a reasonable estimation of the performances of individual unit operations as well as overall plant. In these computations often the mass flow rates and the assay values of various material streams are used. However, in many instances, the streams may not be sampled and therefore the assay data may not be available. Also, the mass flow rate values are not measured for many intermediate streams of a flowsheet. However, with the available data and measurement of minimum additional data may be sufficient for detailed mass balancing and plant evaluation. This is illustrated with an example below. In the Figure 9, the stream name is followed by its identification number.

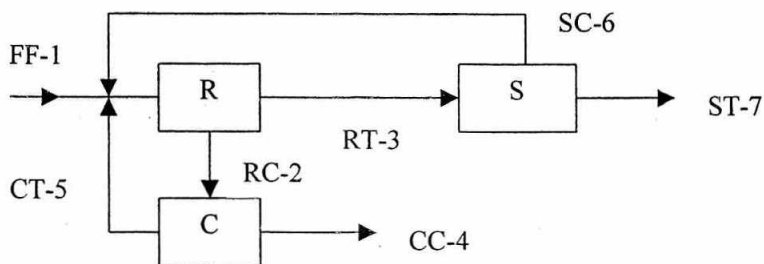


Fig. 9 : A typical circuit for processing in Mineral Processing Plant

In the above Figure, the fresh material is fed to process operation R (stream name FF, ID no 1). The valuable stream from R (RC-2) is further processed in operation C for quality improvement, while the gangue stream from R (RT-3) is processed in operation S for recovering any remaining valuable before discarding. The gangue stream from C (CT-5) is likely to have some valuable and is fed back to process R along with stream FF. The valuable from C (CC-4) is good quality material and is the valuable product of the circuit. Similarly, the valuable stream of operation S (SC-6) is fed back to R and the gangue stream from S (ST-7) indeed has little or no valuable and is the other product (waste) of the circuit.

Usually, the solids mass flow rates of the fresh feed (stream 1) and valuable product (stream 4) are measured. The valuable content (assay) of various streams are also measured. If 'F' denotes the mass flow rate of solids and 'a' denotes the assay value then, say the following data are available (the calculated data are marked so in the Table):

Stream No.	1	2	3	4	5	6	7
Mass flow rate (F, tph)	100	56.94 (calc)	104.21 (calc)	25.82	31.12 (calc)	30.03 (calc)	74.18 (calc)
Assay (a, %)	8.2	16.9	3.6	26.3	9.1	7.8	1.9 (calc)

For the overall circuit the following mass balance may be written:

$$F1 = F4 + F7 \quad \text{for the total mass conservation and}$$

$$F1 \times a1 = F4 \times a4 + F7 \times a7 \quad \text{for the valuable mass conservation}$$

With the data from the Table, the above two equations may be solved to obtain the mass flow rate and assay of the waste product stream ($F7=74.18$ tph and $a7 = 1.9\%$). Thus, overall plant performance may be computed as:

$$\text{Yield} = (\text{mass flow rate of valuable stream})/(\text{mass flow rate of feed stream}) = 25.82\%$$

$$\text{Recovery} = (\text{valuable flow rate in valuable stream})/(\text{valuable flow rate in feed stream}) \\ = (F_4 \times a_4)/(F_1 \times a_1) = 82.81\%$$

Similarly, mass balance around process C gives:

$$F_2 = F_4 + F_5 \quad \text{and} \quad F_2 \times a_2 = F_4 \times a_4 + F_5 \times a_5$$

Solving the above two one gets $F_2 = 56.94$ tph and $F_5 = 31.12$ tph.

So, process C has the following performance indicators:

Yield = 45.35% and Recovery = 70.57%.

Mass balance around process S:

$$F_3 = F_7 + F_6 \quad \text{and} \quad F_3 \times a_3 = F_7 \times a_7 + F_6 \times a_6$$

Solving them for F_3 and F_6 one gets $F_3 = 104.21$ tph and $F_6 = 30.03$ tph.

Yield = 28.82% and Recovery = 62.44%.

Now, the values around process R are known and its performance may be computed:

$$\text{Yield} = (56.94)/(100 + 30.03 + 31.12) = 35.33\%$$

$$\text{Recovery} = (16.9 \times 56.94)/(8.2 \times 100 + 30.03 \times 7.8 + 31.12 \times 9.1) = 71.95\%$$

Thus, the evaluation of individual unit operations and overall plant performance may be done through mass balance techniques.

Modeling Concepts:

The operations in mineral processing can be broadly divided into three categories, namely, transformation, separation and de-watering. The units that change the characteristics of the material are known as transformation units. For example, the crushing and grinding operations change the particle size of the material. The separation units are those that separate the minerals making use of the differences in their properties. For example, flotation unit separates on the basis of hydrophobicity of the mineral surfaces and heavy media cyclones separate the heavy minerals from the lighter ones. Filters and dryers separate the water from the solids. Even thickeners and fine screens are often considered de-watering units for one of the product streams is predominantly water.

This simple classification of operations is effectively exploited in modeling the unit operations and thereby simulating the plant performance. The model of a transformation unit should be able to compute the product characteristics given the details of the feed material along with machine parameters. It may be noted that all transformation units must have only one feed stream and one product stream. Thus in generic form the model of such units may be represented in the vector-matrix form:

$$P = T F \quad \dots\dots\dots(15)$$

Where, P and F are vectors describing the product and the feed characteristics and T is a transformation matrix that defines how the characteristics of the feed are converted into that of the product stream. The determination of this matrix, however, is quite involved and requires in-depth analysis of the mechanisms of the process.

The separation units on the other hand must have a minimum of two product streams. The model of these units should be able to describe the various product streams from the given description of the feed stream. This is achieved by making use of efficiency vectors. The efficiency of transfer of material of certain type to a particular product stream may be expressed in a similar vector-matrix form:

$$P_1 = E_1 F \quad \dots\dots\dots(16)$$

$$P_2 = E_2 F \quad \dots\dots\dots(17)$$

..etc., where, P₁, P₂ ... are vectors describing different product streams, F is the vector describing feed stream and E₁, E₂ are efficiency matrices signifying transfer to a specific stream. It may be noted that the efficiency matrices are all diagonal matrices and they must add up to 100% (Identity matrix). Thus, for a two-product separation process E₁ = I - E₂. Again, the estimation of these efficiency matrices depends on the science of the process. The approach for the estimation of these matrices varies widely with each process and may be deeply involved task. However, such a description of the calculated product streams are then passed to a simulation tool as feed to a subsequent operation in the plant flowsheet. A complete simulation of the entire flowsheet could thus be made possible.

FURTHER READING

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